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METHOD AND COST OF MINING AT NO. 8 MINE, ST. LOUIS SMELTING AND REFINING COMPANY, SOUTHEAST MISSOURI DISTRICT

BY

ROY H. POSTON

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METHOD AND COST OF MINING AT NO. 8 MINE ST. LOUIS SMELTING AND REFINING COMPANY SOUTHEAST MISSOURI LEAD DISTRICT [1]

By Roy H. Poston[2]

INTRODUCTION

This paper sets forth the mode of ore occurrence, characteristics of ore and country rocks, methods of exploration, development, and ore extraction, and the costs of underground operation at No. 8 mine of the St. Louis Smelting and Refining Works of the National Lead Co. located near the southern extremity of the Southeast Missouri lead district. The author acknowledges the assistance of E. T. Campbell, geologist, in the preparation of the paragraphs on the geology of the ores.

HISTORY

The first record of mining in St. Francois County, Mo., concerns more or less shallow semisurface "diggings" which were worked about the year 1742. In 1864, however, the discovery of the disseminated deposits, the present source of lead, was made on property near the town of Bonne Terre. Winslow has estimated the lead-ore production in this county prior to 1869 to be 59,526 tons.[3]

The introduction of the surface diamond drill for exploration purposes in the year 1869 by the St. Joseph Lead Co. was the means of accurately determining the real extent and possibilities of deep mining.

The St. Louis Smelting and Refining Works of the National Lead Co. began operations in the Southeast Missouri lead district in May, 1898. At present this company owns three areas in the district, as indicated on the map (fig. 1) showing a portion of St. Francois County, Mo. On one of these areas, comprising 40 acres of land, is located the No. 8 mine, operations at which are to be described herein.

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[1] The Bureau of Mines will welcome reprinting of this article, but requests that the following footnote acknowledgment be used: "Printed by permission of the Director, U. S. Bureau of Mines. (Not subject to copyright.)"

[2] One of the consulting engineers, U. S. Bureau of Mines.

[3] Winslow, Arthur., Lead and Zinc Deposits. Missouri Geological Survey, vol. 7, p. 531.

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GEOLOGY

The most authoritative source of information on the geology of this district is the "Geology of the Disseminated Lead Deposits of St. Francois and Washington Counties," comprising parts 1 and 2, vol. 9, of the Missouri Bureau of Geology and Mines. These volumes contain several geological cross sections and maps.

Historical Description.- The disseminated ore deposits are found exclusively in the Middle Cambrian rocks in this area. The two formations of this age present in Missouri are the Bonneterre limestone and the Lamotte sandstone, the latter being the oldest sedimentary formation in the State, the age of which is definitely known. Underlying the Lamotte sandstone are pre-Cambrian porphyries and granites, the granite being instrusive into the porphyries over a large area. At some period in pre-Cambrian time and after the intrusion of the granite, this area was subjected to a long erosion interval during which tremendous quantities of granite and porphyry were eroded. Near the close of Middle Cambrian time this area sank rather suddenly beneath the sea until only the highest parts of the pre Cambrian rocks remained. Large quantities of sand, the result of pre-Cambrian weathering, were deposited in the sea adjacent to these land masses to a depth of 300 or 400 feet. After the first sudden drop the area must have gradually settle until only the tops of the highest peaks and ridges remained above water. Due to the change in elevation and the removal of most of the resid sand from the lan mass, the conditions of sedimentation changed and lime beg , be deposited. This lime is called the Bonneterre formation and was laid down conformably upon t Lamotte sandstone; hence the division between the Bonneterre and the Lamotte is n clear cut, and the base of the Bonneterre has several sandy horizons. The Bonneterre formation consists of about 400 feet of highly magnesian limestone, so of which is argillaceous and is interbedded with layers of dark gray and black shale. Near the base of the formation where the shale layers are the most numero is also found considerable glauconite locally called "chlorite." This is not present uniformly but is rather irregular in horizontal extent; its stratigraphic position is just above the sand and usually below the ore, although in some cases there is ore in this glauconitic limestone.

Above the Bonneterre formation lies the Davis shale which is about 160 t 200 feet thick and consists of green and gray shales and argillaceous lime. It lies conformably upon the Bonneterre.

Conformably above the Davis shale is the Derby formation. This is about feet thick and consists entirely of dolomite. It is usually grouped with the ov lying Doerun formation, which is about 50 feet thick and consists of thin-bedded argillaceous dolomite. During the deposition of the Davis and Derby - Doerun fo tions, shallow water conditions prevailed. At the close of Doerun time the area probably rose above the sea. This lift was probably gradual in contrast to the previous sudden drop.

Capping the highest hills around the Flat River district is the Potosi formation, which is several hundred feet thick and is characterized by quantitie of drusy cherts. This formation was very cavernous and open and most of the ope ings have been lined with quartz druses and chalcedony. Upon weathering, the dr

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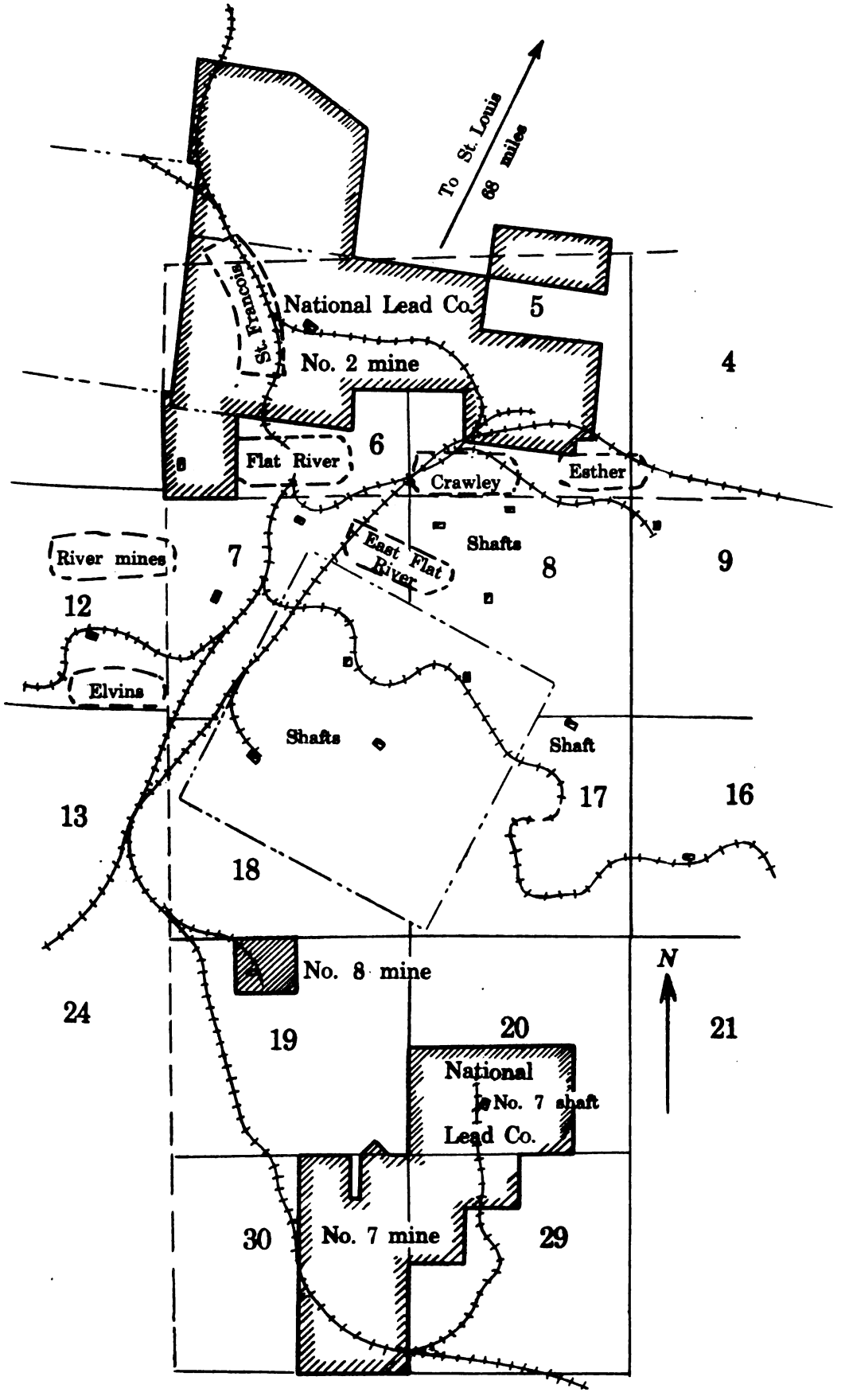

Figure 1. — Key map showing location of mines of National Lead Co. in southeast Missouri

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are left to form a protective cap which greatly retards erosion. This formation does not indicate the presence nor the absence of lead ores, but in diamond drilling it is avoided, if possible, because of the effect of the quartz-lined pockets on a diamond bit.

Structure.- It is not to be expected that such diastrophic movements as have taken place in this area could have occurred without some faulting, and we find between the Flat River district and the pre-Cambrian area a large normal fault, a displacement of about 600 feet. Roughly parallelling this major fault are several minor faults at intervals of a quarter of a mile to a mile. The accompanying pictorial cross section, Figure 2, shows this relationship, as well as the relative position and thickness of the various formations discussed.

Ore Deposits.- The first mines near Flat River were surface diggings for fissure and residual lead. The ore was found in fissures in the rock and in clay openings as the result of the weathering. These surface diggings stimulated interest in deeper ore and about 1888 the first diamond drills were brought to the Flat River area, although they had been introduced at Bonne Terre about 1869. The drilling disclosed large bodies of disseminated ore lying for the most part within 150 feet of the Lamotte sandstone, although this distance is somewhat variable. Since that time much diamond drilling has been done regardless of surface showings and many ore bodies have been discovered without the aid of any lead showings on the surface. In a few instances disseminated ore has been found beneath surface deposits, but many surface deposits have been drilled without finding any ore beneath them. The presence or absence of surface ore has been practically disregarded as a guide in drilling. The disseminated ore bodies near the base of the Bonneterre formation are rather irregular in shape, although they usually have a rather definite trend for each local area of mineralization. For example, the distance from the No. 4 shaft of the St. Louis Smelting & Refining Works to the No. 6 shaft is about a mile and a half. The ore is almost continuous between and for a short distance beyond these shafts and is several hundred feet wide. This ore body strikes roughly northeast and southwest. The ore body in which are located shafts No. 7 and No. 8 strikes roughly northwest and southeast, and is about 3 miles long and from 300 to 800 feet wide. At No. 8 mine, which is the particular property described in this report, the ore is mined at two levels and is only about 35 feet thick. At No. 4 mine there are several places where the stopes are 200 feet high. The ore consists of galena, marcasite, pyrite, and chalcopyrite associated with calcite, disseminated in the dolomite as a filling in small fissures and cavities and in seams in the shale and shaly dolomite. Only the galena occurs in sufficient quantities to be of value. At some places in the district enough zinc occurs with the ores to make a separate zinc concentrate in milling the ore from parts of the mines, but no zinc has been observed in the ore at No. 8 mine.

The effect of faulting on the ore bodies is not very definite. The faults undoubtedly have served to allow the mineralizing solutions free movement, but as a rule the faults themselves are not mineralized. This becomes less true as the throw becomes less. The faulting that has been observed in the mines is rather clean cut with little or no gouge. In some places the movement of the water in these faults has dissolved the limestone, enlarging the openings and somtimes

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filling them with red mud from the surface. In mining it is necessary to be very cautious about drilling into these enlarged faults or fractures, commonly called channels, as the water encountered often constitutes a serious problem. In the past several ponds and streams have drained into the mine workings because of cutting into these channels.

Origin of Ores.- As to the origin of the ores there have been many theori advanced by eminent authorities, among them the following: J. E. Spurr contends that the ores are magmatic injections; W. H. Emmons attributes them to ascending hot waters; H. A. Buehler, who has probably studied the district in the greatest detail, attributes them to gradual deposition by descending cold waters which toc the lead from the overlying rock as it was eroded away. According to his theory the lead was deposited in these overlying rocks during their formation and originated in the pre-Cambrian period. The disintegration of these rocks, and th transportation of the lead in solution by rivers and streams to the surrounding seas where it was precipitated by reaction with organic matter, accounts for the initial deposition of lead in the sediments.

EXPLORATION

In common with many lands adjacent to known mineable areas the No. 8 mine area was first tested with the surface diamond drill to determine whether or not lead ore existed. The first few holes showed that lead ore did exist at depth, and eventually some 80 holes were put down on this property at irregular interval and sufficient data was accumulated upon which to predicate extensive underground development. These 80 holes comprised some 48,000 feet of drilling, of which records have been very carefully kept.

The first few holes were cored from the surface to the sand, but after th approximate ore horizon had been definitely established the first few hundred fe of each subsequent hole were "plugged;" that is, drilled with a center bit, ther by eliminating the core through this section. This procedure saved both time an money without loss of pertinent geological data.

When drilling through the ore horizon, detailed records were kept in the field by a core inspector. This inspector checked the depths of each core run when drilled, suitably marking the cores in the core containers; he looked after sludge samples, checked hole numbers, and stored the cores until they were trans ferred to the assay room at the main plant.

All cores containing lead were assayed to determine the exact lead conte and the results were compared with the assays of the corresponding sludge sample Sludge samples were used solely as a check upon core sample assays, and the latt values were used for all ore-reserve calculations.

The cost of this drilling varied from .95 cents to $1.40 per foot, the variation being due largely to physical conditions beyond the company's control. The drilling was done with equipment owned and operated by the company.

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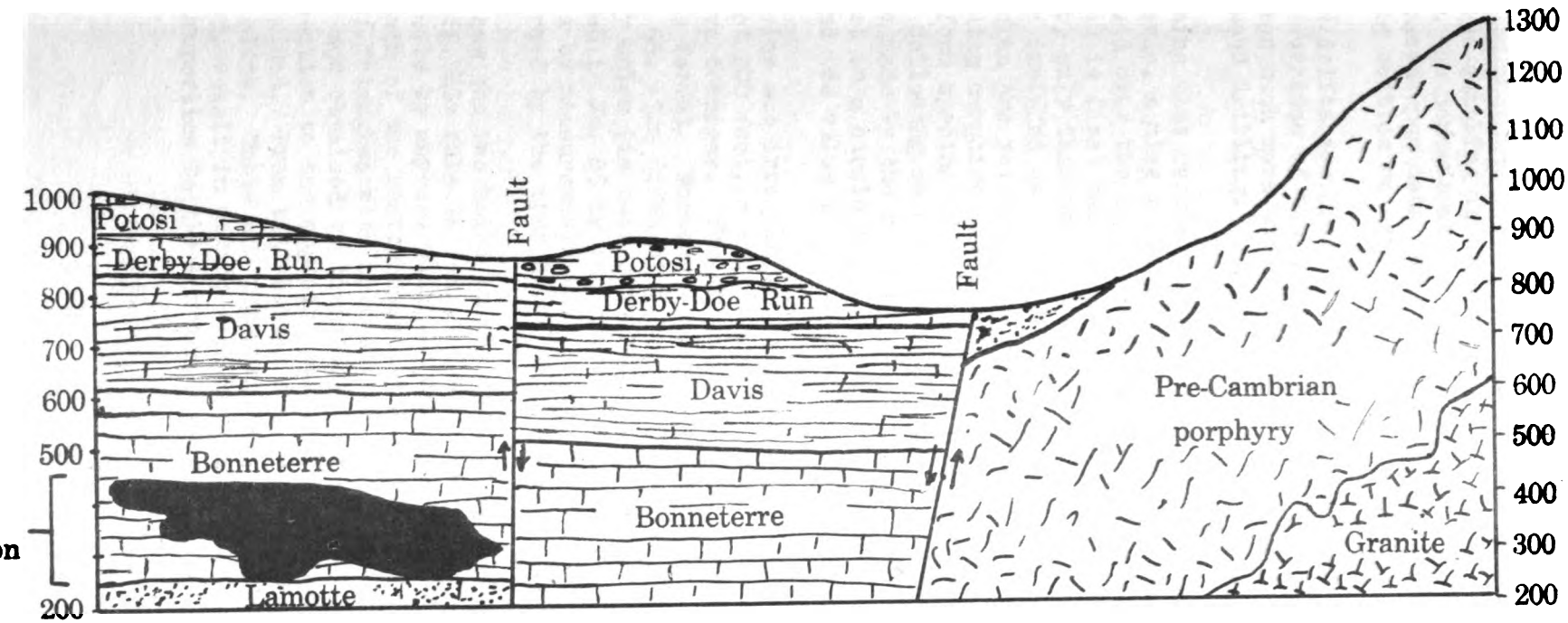

Figure 2. — Pictorial cross section of geological formations from Flat River south to Simms Mountain. No horizontal scale

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METHODS OF SAMPLING AND ESTIMATION OF TONNAGES AND VALUES

No mine or stope sampling is done at the No. 8 mine, but the ore is sampled and assayed at the concentrator separately from that from the other mines. All mining plans are based upon data obtained from the surface drilling, as described in the preceding section on exploration.

No exploration drifts are driven into virgin ground because the irregularity of the occurrence of the ore and its uniformly flat dip would make this form of exploration much more expensive and less effective than the present system of surface diamond drilling.

Ore at No. 8 mine that carries less than 14 feet per cent lead content is not mined; in other words, a mine stope must be at least 7 feet high and must carry at least 2 per cent lead over the entire stope face. Where more than 7 feet are involved, a calculation is first made to determine whether the entire thickness is minable. If it is not, only that portion averaging 2 per cent lead and more than 7 feet in thickness is included as ore. Ore in place is calculated at 12½ cubic feet, per ton, and broken ore weighs approximately 2000 pounds per 20 cubic foot. Before the present mining campaign was undertaken, the entire property was drilled with more or less uniform spacing of holes - not on coordinates. This made it possible to apply the following method of estimating ore reserves for comparison with similar estimates made by the conventional formula. For tonnage estimates, ore is assumed to exist in a circle 150 feet in diameter, using the diamond drill hole as its center, and the value is taken as that of the drill-core sample.

Where drill holes are irregularly spaced and fairly close together, say at a maximum separation of 200 feet, a system of polygonal areas is used for the calculation of ore reserve tonnages. The attached sketch, Figure 3, shows in detail how these polygons are formed. For example, it is desired to outline the polygon around drill hole No. 66. The first step is to draw lines from the selected hole (No. 66) to all nearby holes (in this case holes Nos. 6, 76, 28, 14, 67, and 51). Dotted line from No. 65 to No. 66 is incorrect, since the holes included for this purpose are determined by measurements that establish the shortest diagonals of the trapezoid which is formed by the four holes under consideration.

In each instance the two holes at the ends of the shortest diagonal of the trapezoid are used. If this rule is followed throughout, the polygons can be reproduced around each hole by anyone, with the same results, thus eliminating the personal equation. Each of the nearby holes is then connected with each other; this forms a number of triangles with hole No. 66 at the apex. The center of each of these triangles is now obtained by bisecting each side and drawing a light line from the point of bisection to the opposite apex. This is done from each angle, and the center is obtained. From this center a heavy line is drawn to the center of each of the three sides. These are permanent lines and, when all construction lines have been erased, result in making each hole the center of a polygon, the area of which is in proportion to the spacing of the surrounding holes.

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The polygonal method of estimating ore reserves was applied in this particular case because the spacing of the drilling made it particularly applica The custom in the district is to employ whichever of the conventional methods of ore estimation, as determined by experience, may be best fitted to the particula case in hand.

Polygonal areas are measured by means of a planimeter, and the final equation used to obtain the tonnage figure is as follows:-

$$\frac{\text{Surface area (in feet)} \times \text{Estimated stope height (in feet)}}{12\frac{1}{2} \text{ cubic feet}} = \text{Estimated tons of o}$$

This tonnage is assigned the lead value shown by the core assays from the drill hole around which the polygon is formed. All core samples containing lead visib to the naked eye are assayed to determine the true lead content. The core recor then takes the form illustrated by the following example:

590 to 595 feet, 7.2 per cent by assay.
596 to 597 feet, 1.2 per cent by assay.

Then	590 to 595 = 5 feet x 7.2 per cent =	36.0	feet per cent
	596 to 597 = 1 foot x 1.2 per cent =	1.2	Do.
	595 to 596 = 1 foot x 1.0 per cent =	1.0	Do.
	7 feet	38.2	Do.

or $\frac{38.2 \text{ feet per cent}}{7 \text{ feet}} = 5.46 \text{ per cent.}$

The example shows that the section contains the equivalent of 7 feet of 5.46 per cent ore. It will be noted that the 12 inches in this run between 595 and 596 h been aribitrarily assigned a value of 1 per cent. This is done to compensate partly for the known loss of lead in the core sample while drilling and to make complete calculation.

The lead is much softer than the country rock and usually shows evidence wear where the core is broken in the lead ore run. Experience has demonstrated that the total ore mined is always somewhat in excess of the estimate made prior to mining.

Heavy duty air drills using jointed steel are employed to cut long prosp holes in mined-out stopes prior to their abandonment.[4]

SHAFT SINKING

The completed shaft cross section is 8 by 20 feet in the clear, and the area is utilized as is shown in Figure 4. The shaft is 586 feet deep to the bottom of the sump.

4 Poston, Roy H., Leyner Drill in Underground Prospecting. Eng. and Min. Jour. Press., vol. 118, No. 22, Nov. 29, 1924, pp. 856 and 857.

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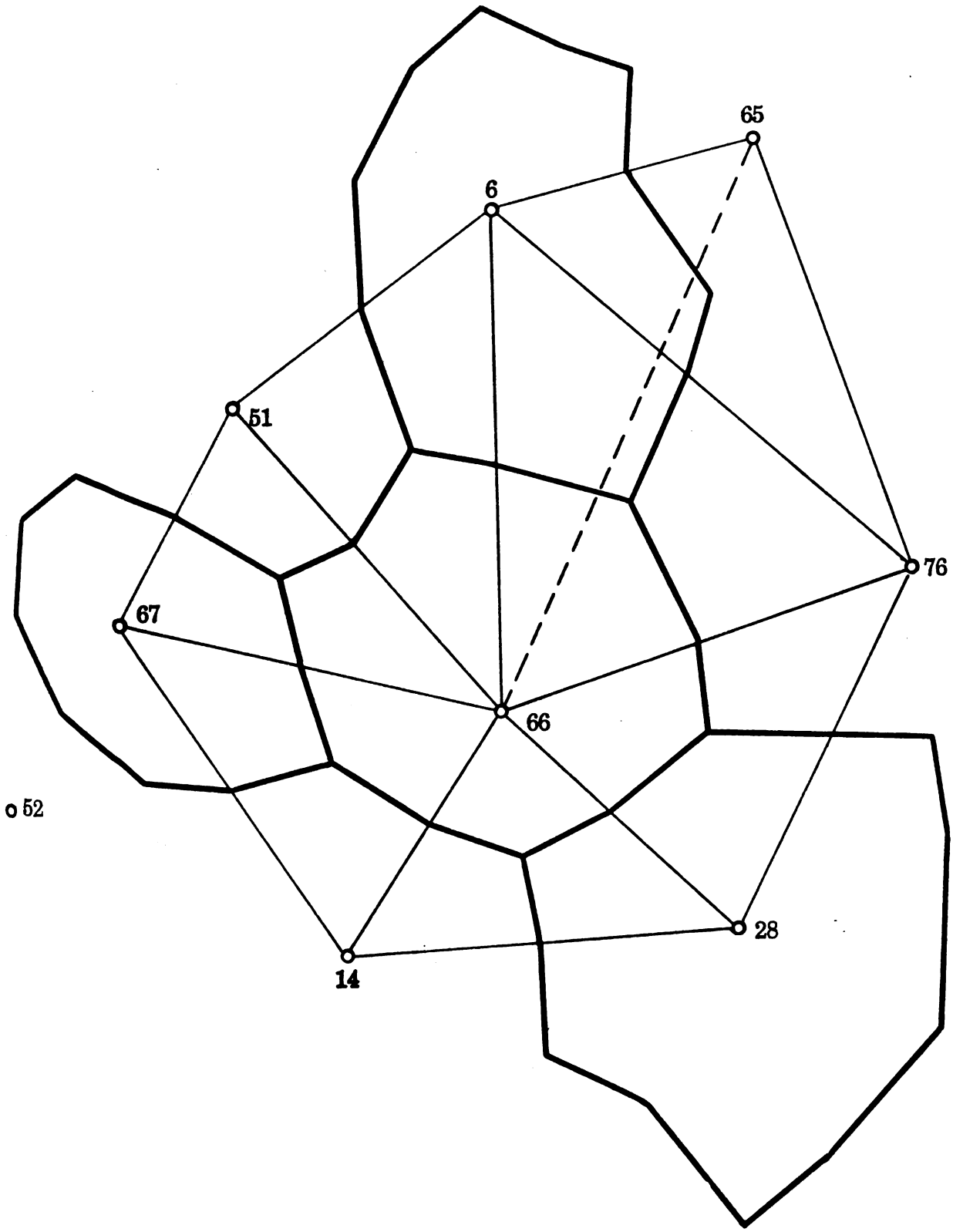

Figure 3. – Method of forming polygonal areas around surface diamond-drill holes

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Actual sinking was done by contractors who were responsible for every phase of the work until the shaft was completed.

The men worked in three eight-hour shifts and the cycle of drilling, blasting, and mucking was followed without regard to the time of day when any particular operation began or ended.

Permanent guide timbers follow the sinking to within approximately 50 feet of the bottom, a stage or "butterfly" serving as a working platform for the timbermen and as a protection for the guides when blasting.

Sixty per cent gelatin dynamite was detonated with electric caps fired by means of a battery.

Drilling was done with jackhammer type drills. A typical shaft round and the order of firing are shown in Figure 5. Holes are filled within 30 inches of the collar, and 7 to 10 cartridges are used per hole.

From 4 to 6 feet were pulled per round and all rock was hoisted by means of a 1,200-pound capacity tub connected to a light-duty electric hoist over a wooden hoisting tower so placed that the final steel tower could be erected around it without interfering with sinking operations.

Air from the drill feed lines was used to clear out the gases after firing so that the men could begin mucking approximately 30 minutes after a round of holes was fired.

EARLY MINING

Early mining at No. 8 mine was prosecuted in much the same manner as at present - that is, with open stopes and pillars, the stopes beginning at the end of development drifts that had been driven to a known ore body as indicated by surface diamond-drill holes.

In the earlier days, however, the difficulties attendant upon a heavy inflow of water of about 1,500 gallons per minute combined with the effect of a low lead market greatly discouraged the development of the mine, so that some little time elapsed between its inception and the present state of more complete development. During this period the inflow of water was greatly diminished by general drainage of the district, and also the surface drilling campaign had more fully disclosed the extent of the ore, so that more permanent equipment was warranted.

The earlier tonnages were handled by use of 1-ton cars on a 24-inch track gauge; the loaded cars were hoisted to the surface by means of a first motion steam hoist, dumped into a small storage bin, and then loaded into standard-guage railroad cars for shipment to a nearby concentrating mill.

A wooden headframe was first used for hoisting, and steam pumps handled the water from an underground pump room until the flow became too heavy for this equipment. Later a vertical centrifugal pump was installed that was belt driven by two single-cylinder steam engines at the surface during this early period of operation. About two acres of stopes and drifts were opened up at this time, and the mine was then allowed to fill up with water and remain idle for several years.

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These two acres of development drifts and stopes greatly simplified the final plans for the resumption of mining at a later date. Figure 6 shows the pla of the layout for the reopening of the mine, and Figure 7 shows in detail the present dumping, storage, and loading facilities at the underground shaft statio

METHOD OF DEVELOPMENT AND MINING

The ore in this mine occurs at two main horizons which overlap on only a very small part of the property. The upper horizon is developed from the shaft a the 560-foot level, which is approximately the depth of the ore below the surfac The lower horizon is about 16 feet deeper and is developed by means of a slope driven on a 4 per cent grade from the upper level, as shown in Figure 8B. Figur 8A is a typical stope section on line A-A of Figure 8B. Cars are handled on thi slope by the mine locomotives.

Development is accomplished by means of drifts and occasional raises.

As the general size, thickness, and depth of the ore body was first dete mined by the surface diamond drill, the shaft bottom was carried to a point sufficiently below the ore body to eliminate any sublevel hoisting through winze and to allow easy grades for motor transportation from headings to the ore pocke at the shaft.

When the shaft had been completed to a depth of 586 feet, three drifts were driven to different points in the ore body to provide points of attack for mining. Standard drifts are approximately 7 by 8 feet in cross section, as is shown in Figure 9. The drift round is also shown in Figure 9, and the order of firing is designated by the small figures near the collar of each hole. The figures in the circles indicate the number of sticks of powder in each hole.

Depths of holes and their direction are also shown in the cross section. 1-1/8 by 8 inch sticks of 35 per cent gelatin dynamite are detonated with a No. detonator placed in the third cartridge from the top.

Fine rock in tamping bags is prepared underground for stemming, and all loading and firing is done by the contract drift driller at the end of each shif The same man handles all misfires.

The depth of cut pulled varies from 3 to 6 feet, depending upon the natu of the ground, skill of the driller, etc. From 14 to 25 tons of rock are broken each round and are mucked out by one or two men, depending on the amount of rock broken. Mucking and drilling are carried on simultaneously.

It is sometimes necessary for the drift driller to shovel some rock back from the heading at the beginning of the shift in order to provide room for the drill column, but in a few hours the shoveler can remove enough of the rock to all him to make his second and last set up.

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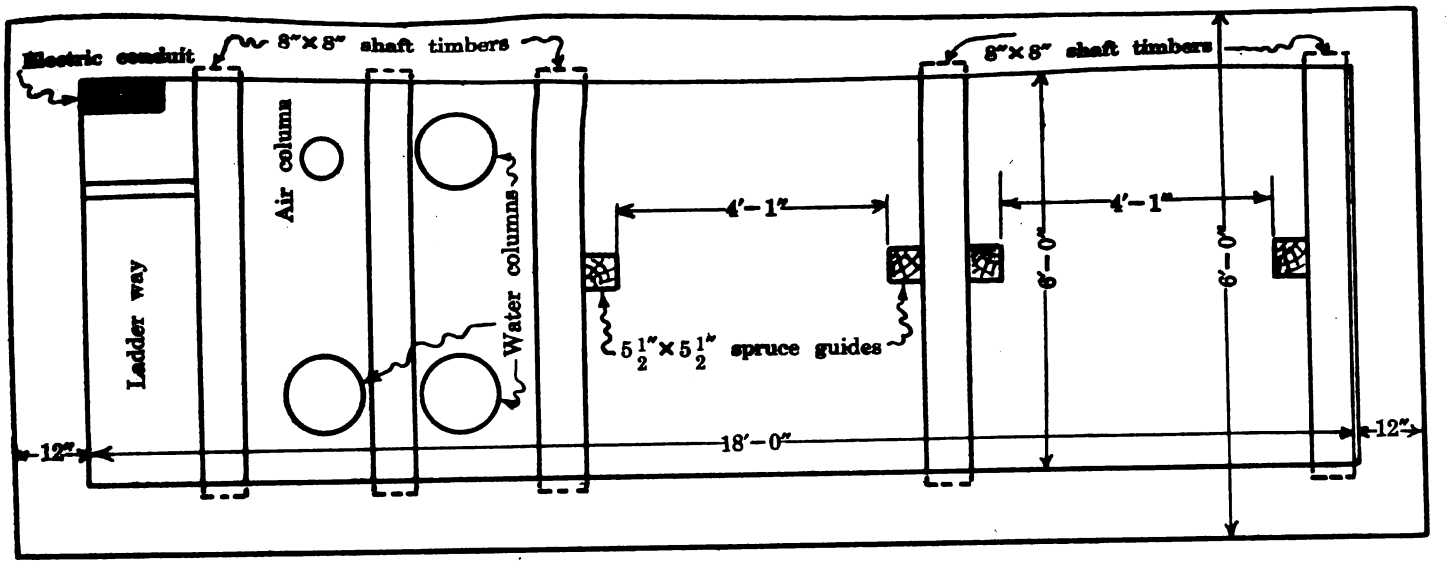

Figure 4. – Plan of shaft

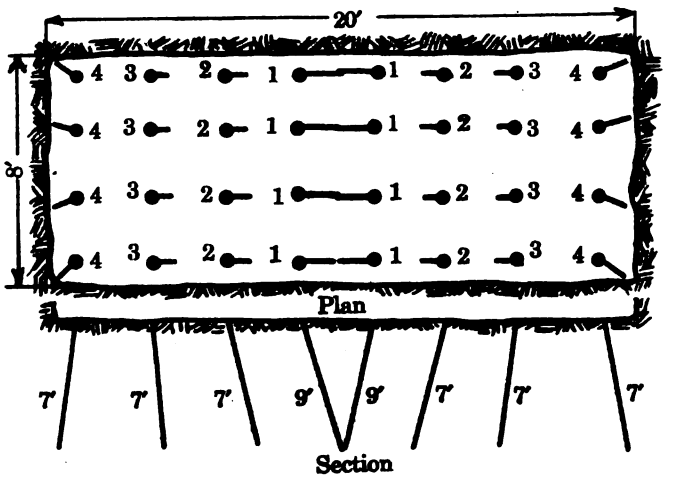

Figure 5. – Typical shaft round

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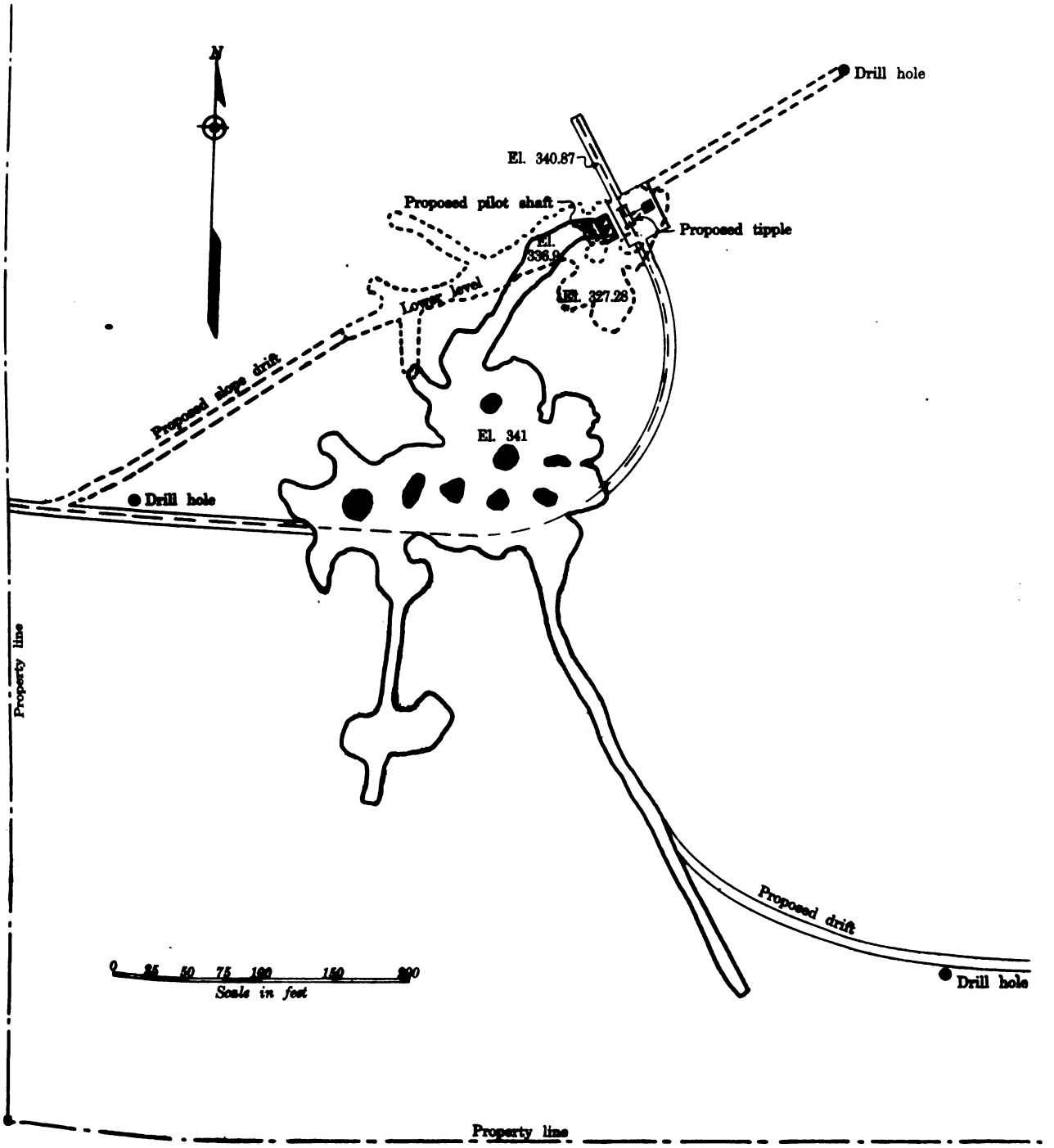

Figure 6.– Plan of layout for reopening No. 8 mine

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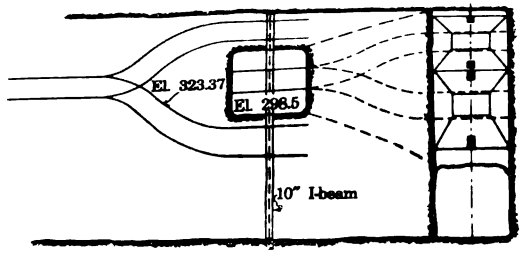

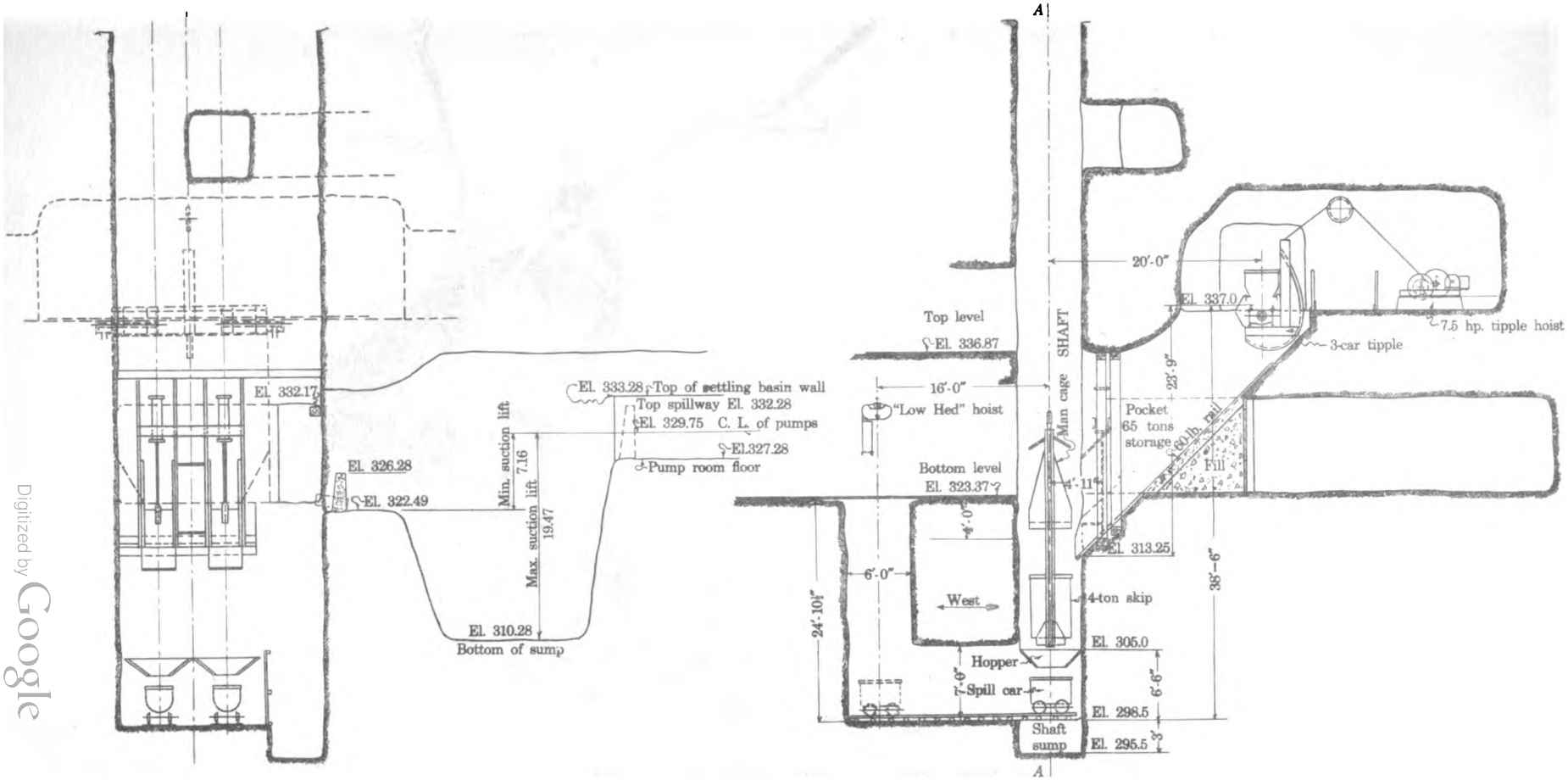

Figure 7.– Dumping, storage and loading facilities at No. 8 mine

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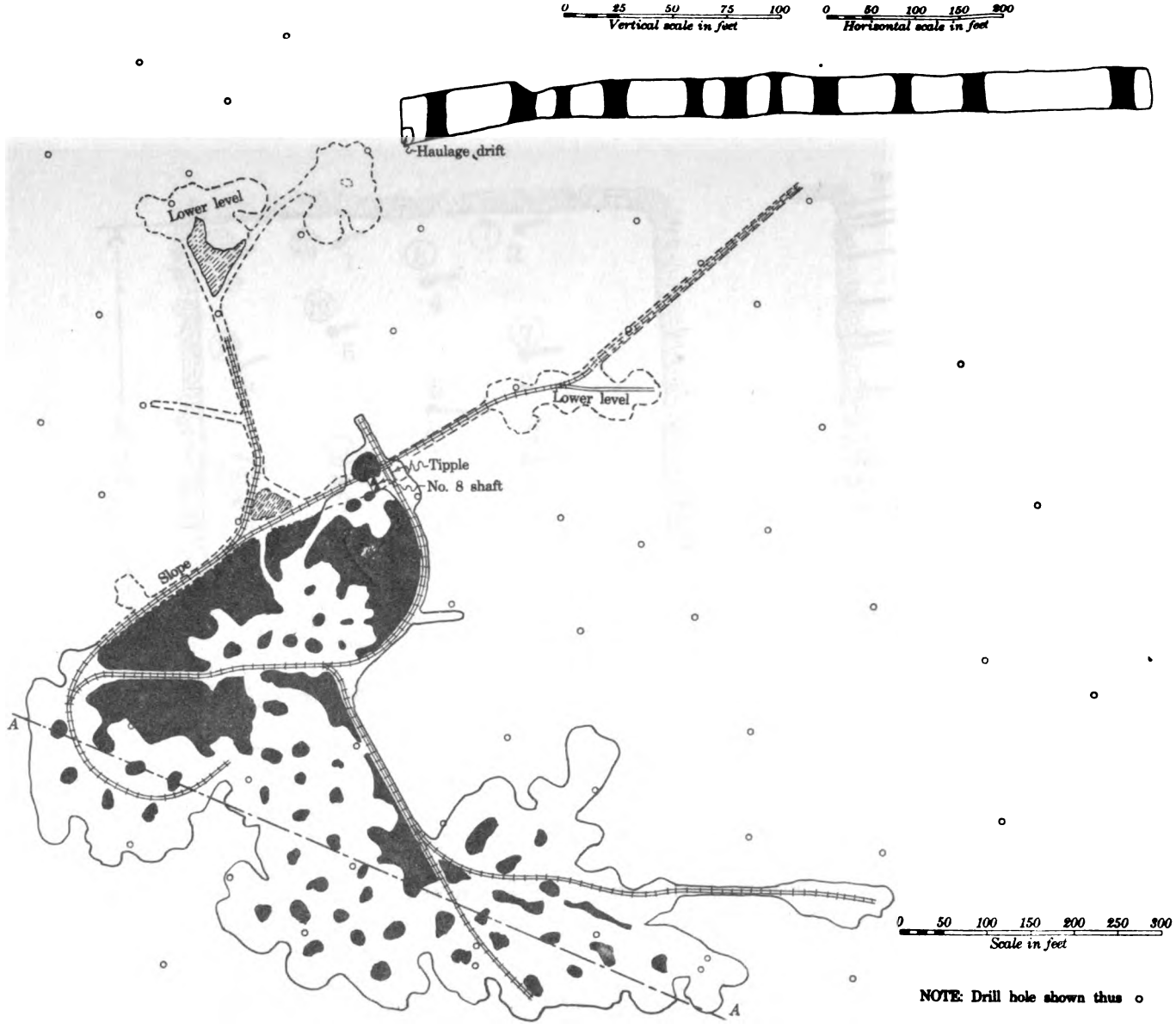

Figure 8.— *A*, Typical stope section *A*–*A*; *B*, plan of mine No. 8

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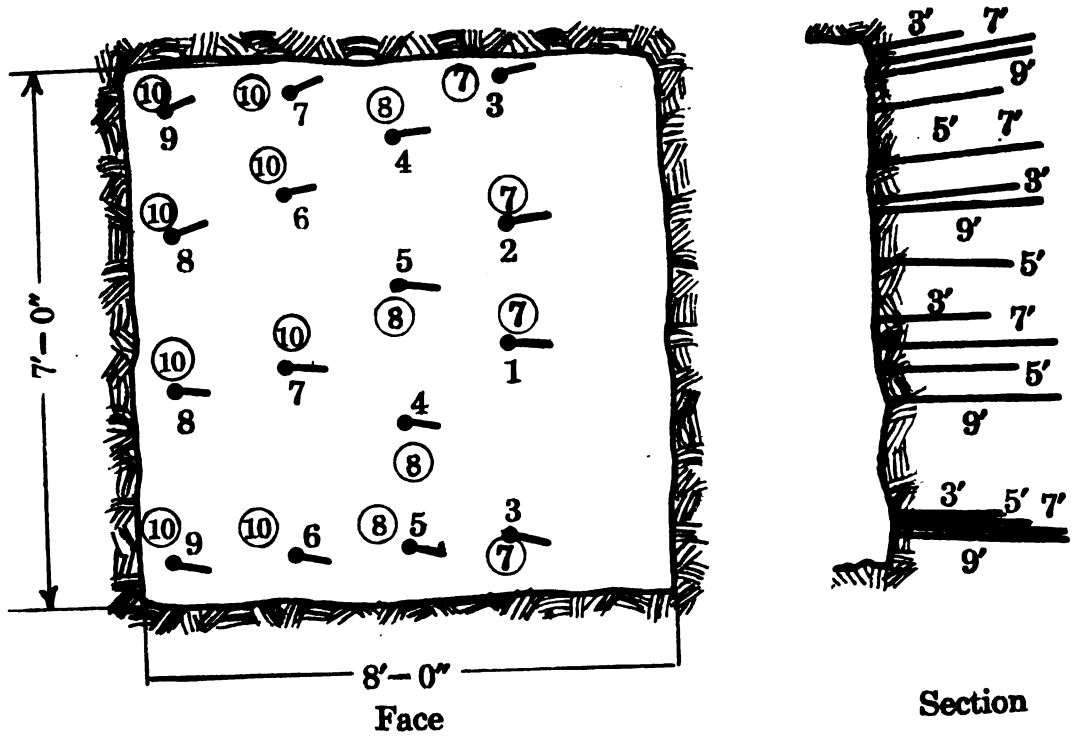

Figure 9.– Typical drift round

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A wet-type jackhammer drill operated by one man, who is the contractor, is employed in drifting. The drill is mounted on a vertical column. The contractor is paid a fixed price per foot of advance for his labor, the company furnishing all equipment, air and explosives, with the exception mentioned below. Usually only one mucker is employed, approximately half of whose wages are paid by the contractor out of the contract price and the other half by the company. Where adverse conditions require that two muckers be employed, the company pays the wages of the second mucker.

Each drift contractor is allowed a maximum of $3.50 per month by the company for drill repair parts. If this amount is exceeded the contractor must pay the excess amount. Ordinarily drill repairs do not exceed the $3.50 set by the company.

In any case the company supplies the labor for inspection and repair of the drill. All drills are tested semiannually for air consumption and cutting speed, and sufficient repairs are made at time of testing to keep them in the proper state of repair. Decision as to necessity for repairs is made by the underground shop foreman, and this decision can not be overruled by the contractor operating the drill in question. This same allowance, inspection and repair system applies to the contract tonnage or stope drills, the difference being that the allowance per drill is $2.50 per month.

Raises and winzes are seldom driven at the No. 8 mine. Most of the lead-bearing lenses are in such position vertically that no raise or winze is necessary to mine them, so that for all practical purposes this mine may be considered a one-level mine.

For mining, two tonnage drills are assigned to each completed drift, one on each side, and in a short time a third drill is added to the original two, making three in each heading; thus three stopes are tributary to each haulage drift.

STOPING METHOD

The ore is mined in open stopes supported by pillars, which are spaced as required for support of the roof. The limestone capping is hard and generally self-supporting over spans of 40 feet, and in some places the span between pillars is as much as 60 feet. In some of the other mines spans as great as 90 feet are self-supporting. Little spalling takes place upon exposure to the air. No timber is employed for ground support.

Figure 8B shows the plan of the mine and pillar spacing. The stopes are driven the width of the area between pillars.

Figures 10 and 11 show the method of stoping. Figure 10 is a section showing operations in thick ore and Figure 11 shows the method of advancing the breast and stope. The ground, which is strong but breaks well, is drilled with jackhammer-type wet drills operated by one man. A heading is carried 6 to 8 feet in height directly under the top of the ore and the stope is benched with down holes as shown in Figure 11. In drilling the headings the drills are either mounted on drill columns or are used with a homemade column as described below.

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The average height of ore is about 20 feet, the maximum being 35 feet.

Holes are drilled to depths varying from 4 to 18 feet, starting with a 1-7/8 inch gauge bit and decreasing one-eighth inch for each 3-foot change in steels. Hollow seven-eights-inch hexagon steel with 1-inch shanks is employed.

Tonnage or stope contractors frequently use a homemade hand column, illustrated in figure 12, for drilling holes in positions where regular columns can not be used to advantage. These hand columns are used to support the weight of the drill while it is in operation.

In use, the hand column is placed in an upright position, and the drill handle is placed over the two spikes at the proper height; the contractor keeps the board in an upright position, pushing it forward with his toe to supply the required feed for the drill.

Stope holes carry 3 to 5 feet of burden and are blasted with 35 per cent gelatin powder. The contract driller handles all drilling and blasting operations, and firing is done at the end of the working shift unless the stope is isolated, in which case firing is permissible at any desired time. In small or congested headings compressed air from the drilling mains is allowed to escape after blasting to facilitate the clearing of the gases.

Comparatively little blockholing is necessary, as the ore breaks well and usually has enough cleavage planes to allow some breakage with sledge hammers. The entire cycle of drilling, blasting, and mucking is completed in each working shift.

Each tonnage drill contractor is allotted a stope and is responsible for the safety of the roof and pillars therein; no roofmen are employed except for unusual conditions. General instructions for each day's drilling are usually issued by the mine captain or shift boss, but the details of operation are optional with the contractor. The company furnishes all supplies and equipment necessary for the driller. He drills, loads, and fires his own holes, and receives a predetermined rate per ton of broken rock, which varies according to the height of the stope.

Muckers or shovelers, both hand and mechanical loader operators, work in pairs on a tonnage basis under the supervision of one assistant boss who is responsible for all these operations, for transportation of the ore to the shaft, and for necessary repairs and renewals to the haulage track and equipment.

The use of so-called bulk dynamite for breaking operations was given a thorough trial at No. 8 mine during 1928 and met with marked success. A bulk dynamite with a stick count of approximately 170 sticks per 50-pound box was used, together with some 35 per cent ammonia gelatin.

At present the percentage of gelatin and bulk dynamite used for breaking is as follows:

75 per cent bulk,
25 per cent ammonia gelatin 35 percent of strength.

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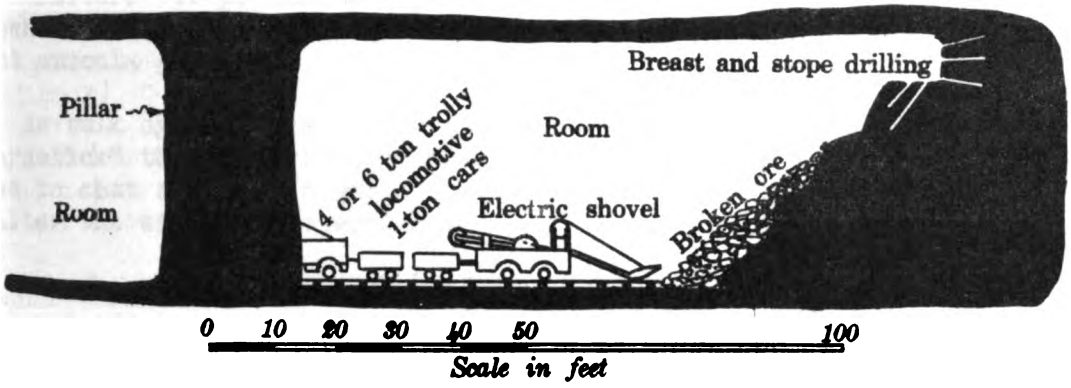

Figure 10.– Section showing operations in thick ore

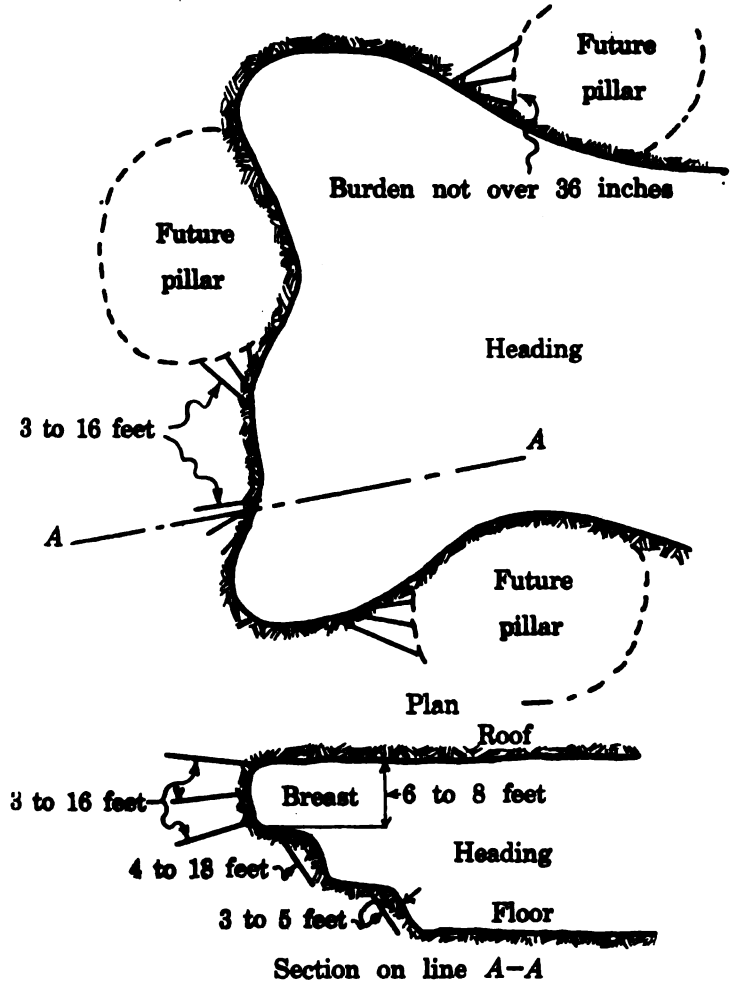

Figure 11.– Method of advancing breast and stope. Not drawn to scale

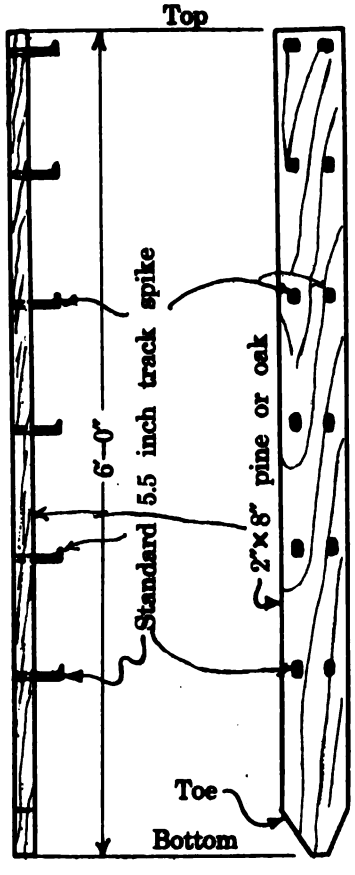

Figure 12.– Home made hand drill column

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"Nifters" or primers with No. 6 caps are also made of bulk dynamite, although when bulk dynamite was first used it was customary to make nifters of 35 per cent ammonia gelatin.

As bulk dynamite is loaded stick for stick when replacing gelatin dynamite, the "yardstick" that must be used when price changes occur is cost per stick. A decrease in cost of gelatin and a corresponding increase in cost of bulk dynamite might alter the situation overnight.

UNDERGROUND LOADING AND TRANSPORTATION

Comparatively little ore is drawn through chutes and raises, excepting, of course, through the skip loading pocket and its chute at the shaft, which are shown in Figure 7.

Four-ton trolley haulage motors, powered by 250-volt d.c. double-unit motors, direct geared to the drivers, are used for gathering and transportation to the shaft skip pocket. These motors travel over 30-pound rails with 24-inch track gauge and receive current through a trolley pole and the grounded rail system. One motorman and one switchman constitute the operating crew.

Track grades are kept below 4.0 per cent, and thus the motor is able to handle from 8 to 14 loaded cars per trip. The motor delivers the empty 1-ton cars to a by-pass or dead-end track, usually located within a few yards of the muck pile.

The loading track is usually constructed with a small grade in favor of the loaded cars. The hand shovelers tram the cars from the empty track to the required position for loading, and then back to the full track when loaded. This is the only hand-tramming done in the mine. The 2-ton cars are spotted by the motor for loading and again removed when loaded.

A mechanical loader is employed in No. 8 mine where tonnage is sufficiently large to warrant its use. This machine is mounted on the standard 24-inch guage mine track, and the ore is loaded by forcing the digging element into the muck pile by the tractive effort of the machine. The dipper delivers its load onto a conveyor belt which transfers it to the car. (Figure 10.)

The car to be loaded is attached to the rear end of the loader by means of a short chain, and the car and loader move back and forth along the track while the car is being loaded. One operator and one helper, who tram the cars by hand, constitute the crew.

Ore is dumped directly into the skip pocket at the shaft by means of a rotary dump or tipple which handles three 1-ton or one 2-ton cars at a time (fig. 13). This tipple dumps the loaded cars by gravity by releasing the brake on the drum of the hoist. The hoist, which is operated by a 7½-hp. motor, is employed to return the dump to the upright position. As the tipple rotates, the rope travels in the curved channel shown in the illustration. Figure 7 shows the assembly and positions of tipple and hoist.

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The dump operator maintains a count on the incoming cars, which must check with the motorman's and the assistant shift bosses' count for each shoveler concerned.

The ore is drawn directly from the main pocket into a 4-ton skip by a skip tender (figure 7). A standard 10-ton wagon scale is used underground to check the weight of the ore loaded before it is dumped into the chute. The cars, which are permanently numbered to prevent confusion, are weighed, loaded, and again weighed after dumping. Car numbering is of further use in determining the life of each car and its repair and maintenance cost.

A 6 by 6 foot pilot shaft, the center line of which is 16 feet west of the center line of the main shaft, gives egress through a short tunnel from the bottom of the skip pocket and provides opportunity to muck out spillage from the skips. Standard 1-ton cars are used on 24-inch gauge track for gathering this spillage, and an air hoist raises the loaded car up through the pilot shaft to be taken to the rotary dump at the shaft.

Four-ton skips attached to 1¼-inch cables are hoisted through the 568 feet of shaft by a single drum hoist and are dumped into a wooden ore-storage bin that feeds through metal chutes directly into standard railroad cars for shipment to the main plant for milling. Details of hoisting equipment are as follows:

The shaft hoist is a single, rigid drum type with single reduction gear, the pinion running between two bearings and connected to the motor by Francke coupling, and was built to the following specifications:

Weight of skip	9,000 pounds
Weight of ore	8,000 pounds
Present skip travel	618 feet
Maximum skip travel	700 feet
Rope speed	750 feet per minute.
Size of rope	1¼ inches

Operated as a balanced hoist.
Drum shaft bearing, 8 inches in diameter by 15 inches long.
Pinion shaft bearing, 5 inches in diameter by 15 inches long.
Drum cast iron turned and grooved for 1¼ inches rope, 72 inches in diameter by 66-inch face.

The gears are of cut steel and of the herringbone type approximately 94 inches in diameter by 12-inch face. The gears are shrouded by guards.

The brake is a heavy compound post brake lined with wood and operated by hand; the brake seat is 84 inches in diameter by 8-inch face.

The hoist is equipped with a "Lilly" controller driven from the main shaf by means of gears and so arranged as to break an electric circuit upon overspeedi

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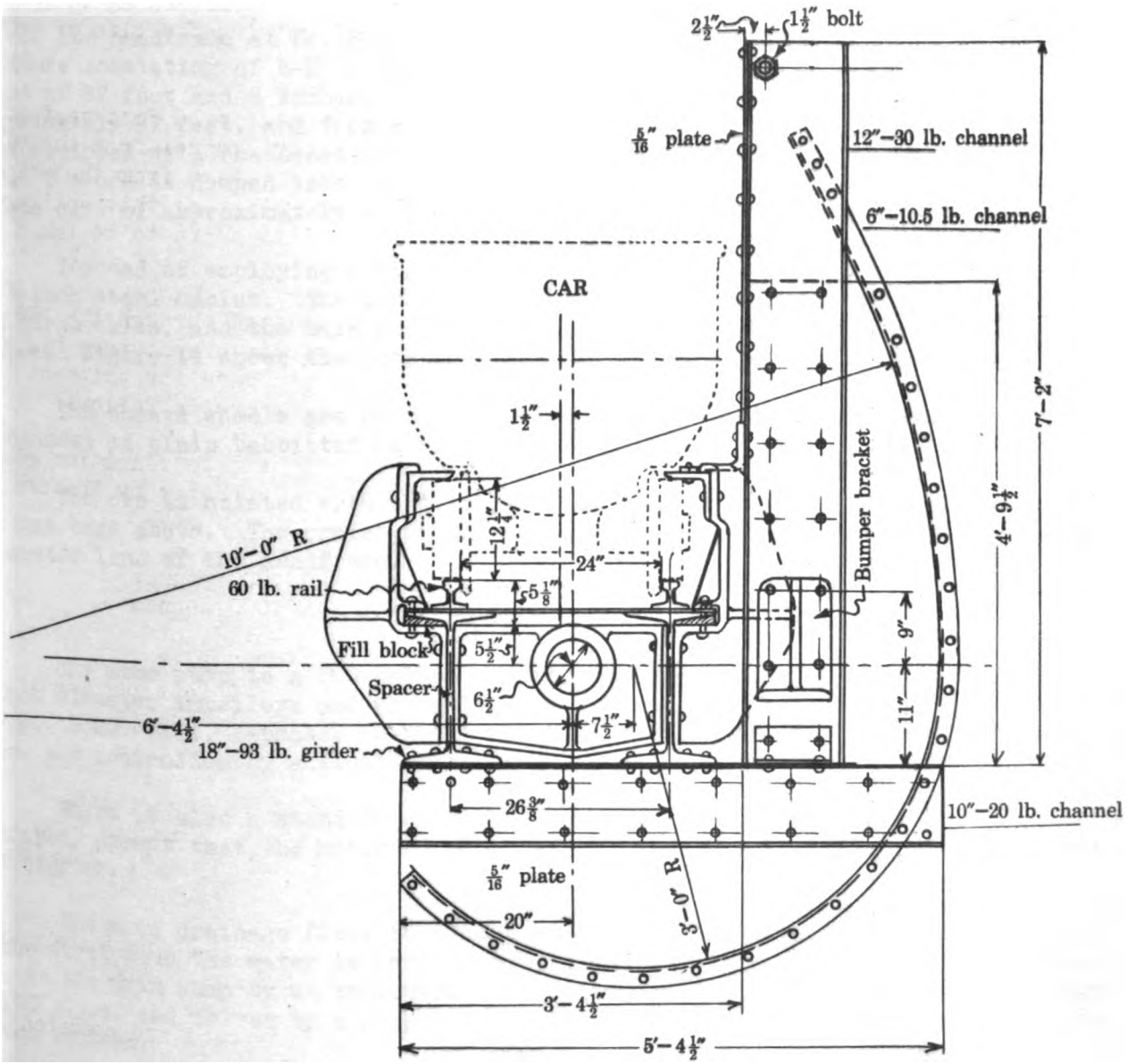

Figure 13.– End view of car tipple

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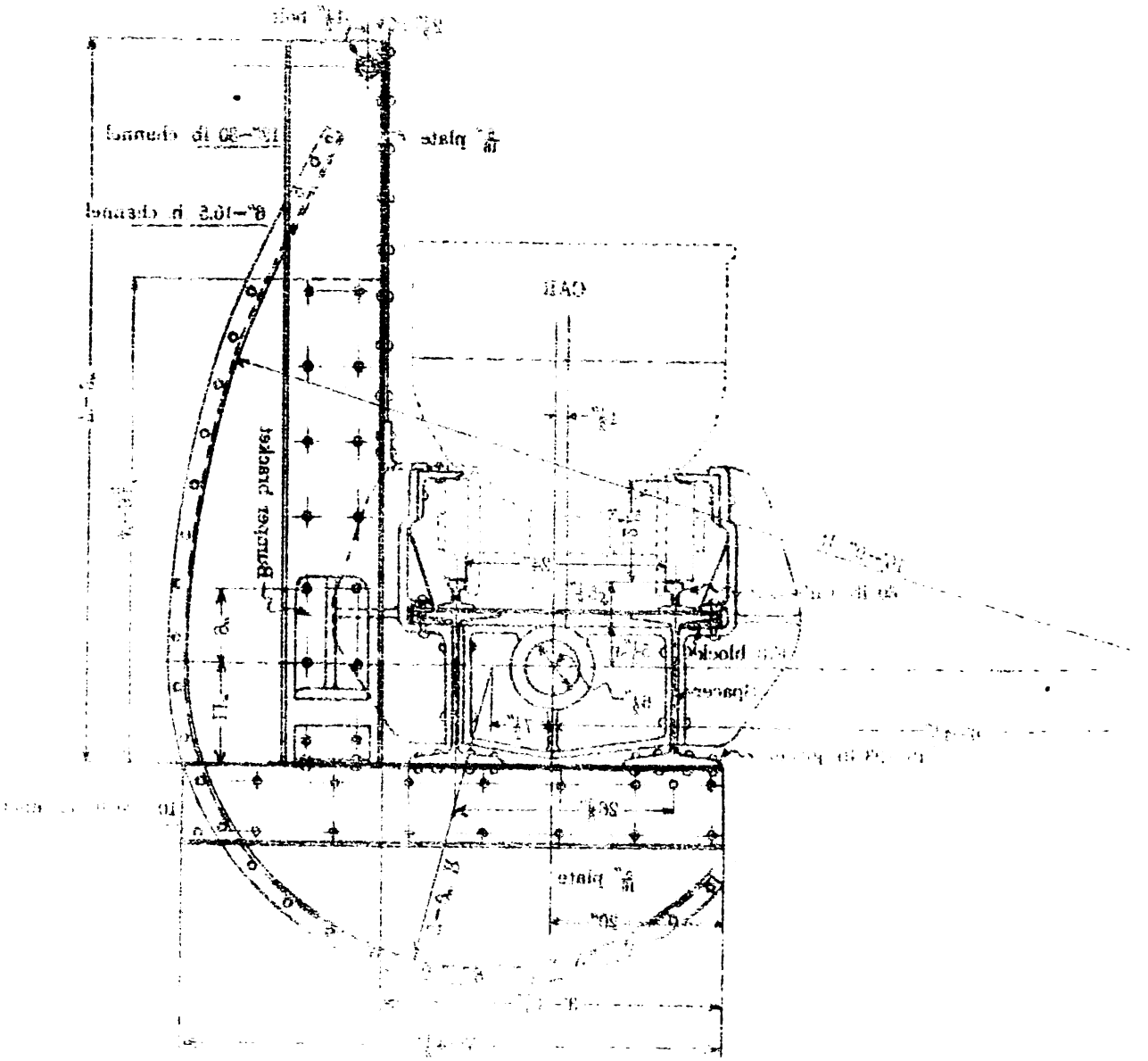

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when approaching the dump, when overwinding, or when starting the hoist in the wrong direction. The breaking of this circuit is arranged to shut off the power and apply the brake, regardless of the position of the operating levers. The brake is applied by weights which are reset by hand.

The hoist is also equipped with a 4-foot dial indicator driven from the drum shaft by means of gears. A platform with corrugated tread on which is mounted the operating levers and controller is an integral part of the hoist. The motor for driving this hoist is a 250-hp., 2,200-volt, 60-cycle, 3-phase alternating current motor, type C.W. 40° C., with wound rotor, 435 r.p.m. at full load.

The headframe at No. 8 mine is a two-compartment, rectangular, steel structure consisting of 6-H beams for main posts with cross bracing, of a total height of 97 feet and 8 inches; from collar of shaft to center line of sheaves is approximately 87 feet, and from shaft collar to dumping point approximately 35 feet. It is equipped with the necessary plate and rails to provide for a self-dumping skip. The ore is dumped into storage bins from which it is drawn into standard surface cars of approximately 40 tons capacity.

Instead of employing a back brace, this headframe is guyed with twenty 1-3/8-inch steel cables. The upper guy lines are held rigid by sag lines provided with turnbuckles, and the main guys are provided with especially designed turnbuckles. Figure 14 shows the details of guy-line bracing for the headframe.

The sheave wheels are of the bicycle pattern, 96 inches in diameter, and are mounted in plain babbitted bearings.

The ore is hoisted with a 4-ton self-dumping skip equipped with a single-deck man cage above. The center of the hoist drum is approximately 165 feet from the center line of the headframe.

PUMPING

The mine pump is a 500-gallon 4-stage turbine type of pump equipped with 13-inch diameter impellers and directly driven through a flexible coupling by a 150-hp., 2,200-volt, 3-phase, 60-cycle 40° C. induction motor, operating at 1,750 r.p.m. and controlled by a float switch and 3-phase 2,200-volt automatic starter.

There is also a stand-by unit which is a duplicate of the one just described, except that the motor is manually operated through a 3-phase 2,200-volt hand starter.

The main drainage flows to a sump several hundred feet from the main sump. At the first sump the water is partly settled and pumped to a settling basin adjacent to the main sump by an enclosed, self-oiling gathering pump having a capacity of 100 g.p.m. and driven by a 5-hp., 220-volt, 60-cycle, 3-phase motor with float-switch control.

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This mine is also equipped with a vertical centrifugal sump pump of approximately 200-gallon capacity and driven by a 5-hp., 220-volt, 60-cycle, 3-phase vertical motor with float-switch control, and is known as the "shaft pump." This pump is placed in the pilot shaft on the lower level of the mine adjacent to the main shaft, and pumps shaft drippage to a settling basin adjacent to the main sump. Figure 15 shows the shaft pump installation.

The mine water is pumped to the surface and discharged into a pond so located that the water from it flows by gravity through the cooling jacket of the air compressor; it also flows by gravity to the change room, where it is used for heating purposes as well as for showers, and for sanitary purposes.

The piping on the surface is arranged in such a manner that by opening one valve and closing another the water can be discharged through a permanent concrete weir box to be used when testing mine pumps.

Power is generated at the power house, approximately 4½ miles away, by turbine-driven alternating current generators producing 6,600-volt, 60-cycle, 3-phase current. The transmission line is bare solid copper supported on chestnut poles on a cleared right of way and transmits the 6,600-volt current to an outdoor substation where transformers are installed to step the current down to 2,300 volts for driving the air compressor, hoist motor, main mine pumps, and motor generator set to supply 250-volt direct current for the underground trolley haulage system. There is also one bank of transformers for stepping the 2,300 volts down to 220 volts for operating the auxiliary equipment, as well as the surface lights and shaft signal; all circuits at the mine are equipped with integrating watt-hour meters that are checked with a totalizing meter at the power house.

PERCENTAGE OF EXTRACTION

As explained elsewhere, stopes that contain less than 14 feet per cent of ore are not considered minable at this time. From the inception of mining to January 1, 1929, approximately 12 per cent of the ground has been left in place as pillars. However, when a lower grade of ore is encountered in an active stope, it is mined along with the other ore on the theory that since the organization and equipment are at hand it is cheaper to mine at that moment than at some future date.

No attempt is made to sort waste rock, as comparatively all rock in the minable area carries some lead.

Practically all rock broken is loaded before the stope is abandoned, so that there are no obstacles in the way of future mining from or through a completed stope.

Careful records to date have shown that in the area mined, 59.3 per cent more tonnage has been recovered than was shown in the ore reserve calculations, or that the experience factor is 1.593.

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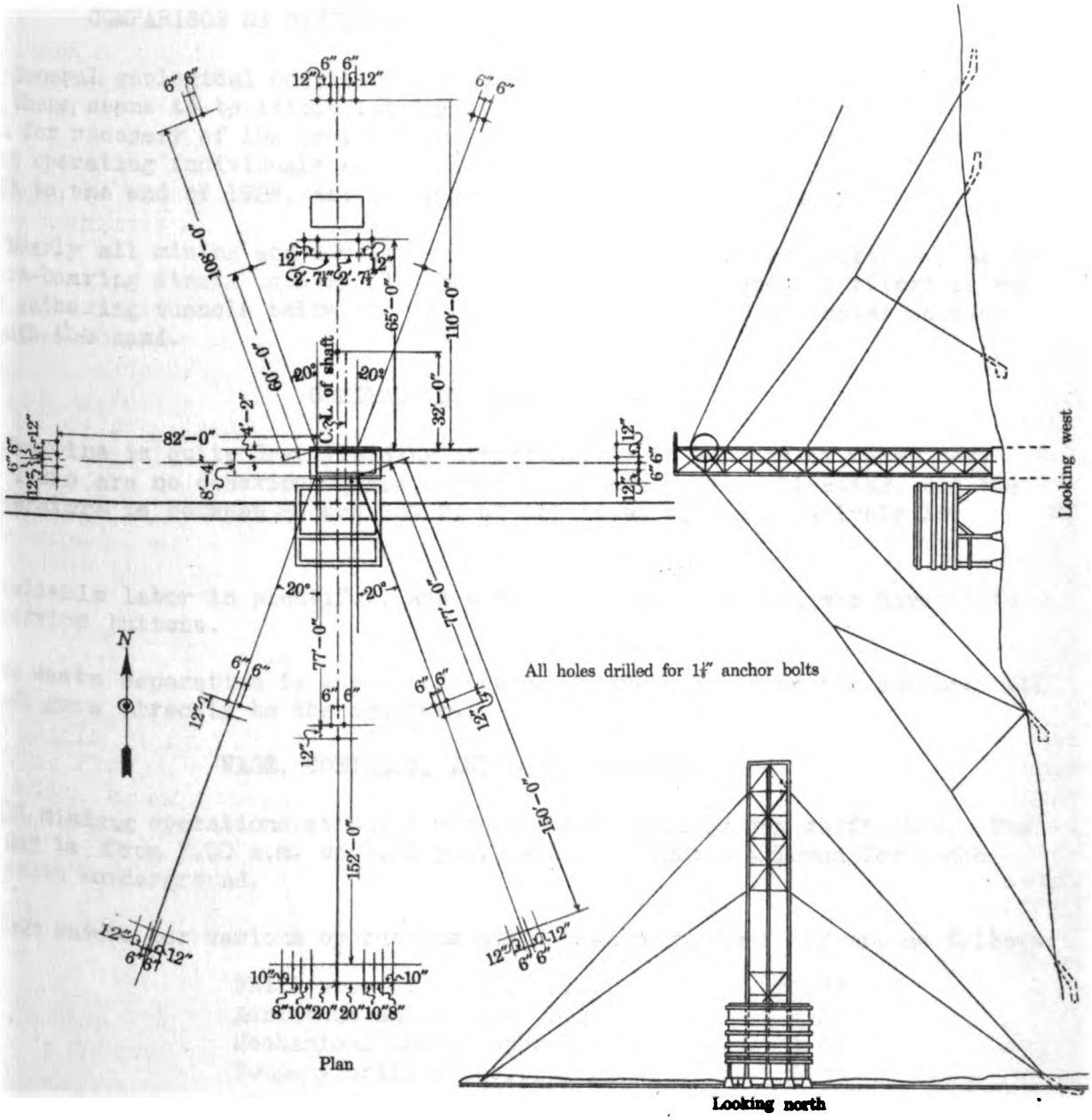

Figure 14.— Head frame and guy lines at No. 8 shaft

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The amount of lead produced in this area was 30.2 per cent in excess of the original calculations; an experience factor of 1.302 has been indicated. It is expected that these experience factors will approach unity as the mine grows older, since the existing workings are in the richest part of the known ore body and because subsequent mining of some of the thinner ore-bearing sections will probably yield less than the calculated tonnages and lead content.

COMPARISON OF DIFFERENT METHODS USED IN THE DISTRICT

General geological conditions are fairly uniform throughout the district, so that there seems to be little latitude as to choice of general methods to be employed for recovery of the ore. It may or may not be significant that the different operating individuals and companies had produced 4,447,000 tons of lead from 1868 to the end of 1928, all using practically the same mining methods.

Nearly all mining and development work is done in the ore horizon. As the lowest ore-bearing strata usually lies on or near the sandstone, no effort is made to drive gathering tunnels below the ore level, since most of the water courses are through the sand.

GENERAL INFORMATION

The mine is quite dry, yielding approximately 75 gallons of water per minute. There are no obnoxious gases except immediately after blasting, and the room temperature is between 60 and 65° F. at all times with a relatively low humidity.

Reliable labor is plentiful, and several men are employed who have 15 to 25 year service buttons.

No waste separation is attempted either underground or on the surface; all broken rock goes directly to the crushers.

WAGE, CONTRACT, AND BONUS SYSTEMS

All mining operations at No. 8 mine are conducted on day shift only. The standard day is from 7.00 a.m. to 3.30 p.m., with 30 minutes allowed for lunch which is eaten underground.

Base rates for various operations and for an eight-hour day are as follows:

Shift boss	$5.95
Assistant shift boss	5.35
Mechanical shovel operator	5.80
Company driller	5.05
Motorman	5.05
Hoisting engineer	5.05
Skip loader	5.05
Hand shoveler (minimum of 18 tons)	5.00
Railroader	5.00
Pumpman	5.00
Switchman	5.00
Pipe fitter	5.00
General underground labor	5.00
General surface labor	3.85

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The rate of $5.80 for mechanical shovel operator applies only when conditions for mechanical loading are very adverse. The operator usually receives 8 cents per ton, and if the earnings at this rate exceed $60 per week, half of this excess is turned back to the company. This refund feature does not apply to the hand shovelers who retain all they may earn under the following schedule:

Contract hand shovelers receive $5.00 for loading 21 tons,
5.55 for loading 22 tons, and
0.28 for each additional ton above 22 to

Company hand shovelers receive $5.00 for loading 18 tons, and
4.50 for loading less than 18 tons, provided the total is approved by the shift boss.

The shift boss's approval simply indicates that conditions were so unfavo able that the required 18 tons could not be loaded.

All breaking is done under a contract system. Each tonnage contractor is guaranteed the company rate of $5.05 per shift. This wage is seldom paid, howeve for the contractors make more when paid on the following basis:

For stopes 8 feet or less in height, 14 cents per ton of rock broken.
For stopes from 8 to 20 feet in height, 13 cents per ton of rock broken.
For stopes more than 20 feet in height, 10½ cents per ton of rock broken.

The rates just quoted are for places where the rock is to be loaded by ha shovelers. When the rock is loaded by mechanical shovels the contractor receives 1 cent less per ton for all rock loaded.

The breaking and loading of all tonnage, exclusive of drift tonnage, is paid for according to actual underground scale weights and not by car units.

Drift contractors receive approximately $4.25 per foot of drift advance, from which is deducted a certain percentage of shovelers' labor necessary to remove the broken rock from the drift. Fifty per cent of the drift contractor's earnings in excess of $72 per week is refunded to the company. Fifty per cent o the tonnage contractor's earnings in excess of $60 per week is also refunded to the company.

Wages are paid weekly by check on each Wednesday for all time worked to 7:00 a.m. of the previous Sunday.

All employees must be ready for duty at 7.00 a.m., but the contract shovelers may leave the mine at any time after they have done the minimum amount of work required by the company. All other employees end their shift at 3.30 p.

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VENTILATION

Ventilation at No. 8 mine is natural. No forced ventilation is necessary since there are no obnoxious or injurious gases other than those released by the daily blasting, which have ample time to be dissipated and escape between shifts.

The use of compressed air for the operation of all rock drills is of course a very important factor in keeping the mine air good, and the fact that one stope in No. 8 mine has been broken into an adjoining mine that is quite large in area assists in providing ample fresh air at all times. The shaft is normally upcast.

FIRE HAZARDS

Fire hazards are practically nonexistent; no inflammable gases are present and there is no timber in the mine with the exception of shift bosses' and mine-tool storage huts, work benches, etc. Hand tank fire extinguishers are placed at certain strategic points, however, such as the pump room, powder storage magazines, mine-tool storage bins, etc.

SAFETY ORGANIZATION

All plant safety work is directed by the local manager and supervised by the plant safety engineer. The chief safety activities consist of various educational campaigns conducted by the several committees.

All plant department heads, foremen, and subforemen, comprise an executive safety committee presided over by the local manager. The duties of this committee are to formulate new safety programs and ideas, discuss former accidents and the moral to be gained therefrom, and to infuse new safety ideas and ideals into the men under their supervision.

This committee meets regularly once each month; oftener, if deemed necessary. Meetings are held on company time at the central office building and are so arranged and conducted that they are not long or tiresome.

The Workmen's Safety Committee, as the name implies, is composed of men chosen from the mine employees. This committee serves for four consecutive bi-monthly meetings. At their last meeting the members select five other men to succeed them for another term of four meetings, and the committee is perpetuated in this manner. The safety engineer meets with this committee, participating in the discussions and keeping suitable records of the sessions. The particular function of this group is to observe, report, and discuss any unsafe condition or practice in the mine, to discuss causes of previous accidents, and to devise methods for prevention of their recurrence. The committee meets and functions on company time, and reports of the action of the meetings, from which many good ideas have emanated, are posted on the bulletin boards throughout the plant.

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The company maintains its own hospital, physician, and nurse in a centrally located building at the main plant. The hospital has an X-ray machine and a well-equipped operating room. First-aid kits are kept at all centers of work activity.

Due to the proximity of the hospital to the main plant, first-aid kits contain only such supplies as are essential when necessary to give first aid. Th contain cotton, adhesive tape, 1, 2, and 3 inch bandages, aromatic spirits of ammonia, a tourniquet, mercurochrome, 1-inch mercuro-compresses, applicators, unguentine, and wooden splints. On top of each kit is a sealed envelope containing the key to the kit. An injured man is requested to open the kit, use necessary supplies, make notations on envelope and then lock the kit, put the key in a new envelope, and place it on top of the kit. These envelopes are gathered every third day.

Periodical training in first-aid methods and resuscitation has been given to all foremen, and educational campaigns are constantly in progress.

Bulletin boards located at strategic points both on the surface and under ground have displayed thereon many items of interest as well as the regular posters from the National Safety Council, The Elliott Service Co., Mather and Co. and occasional homemade posters on some local condition or accident.

Use of various types of eye goggles have materially lessened the number of eye injuries.

Safety types of shoes have been tried in various parts of the plant with indifferent success.

All employees are given a thorough physical examination each year.

Complete and detailed records of all accidents and first-aid cases are kept and systematically analyzed and filed for reference. From these data an acc dent record covering the men under each foreman is compiled and compared with the records of other foremen.

In 1926 the plant as a unit received honorable mention in the National Safety Competition for the "Sentinels of Safety" trophy, sponsored by the U. S. Bureau of Mines. Since that year the various mines have been entered as single units instead of as a whole, on the theory that this arrangement will stimulate the men in each unit to greater efforts. Table 1 gives accident data concerning No. 8 mine for 1928.

An industrial eyesight survey was conducted by the Kindy Optical Co., St. Louis, Mo., for the St. Louis Smelting & Refining Works of the National Lead Co. in the spring of 1928.

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Survey data given here is for the entire St. Francois plant.

	Number	Per cent
Employees who did not need glasses	511	59.9
Employees who were wearing glasses that were not satisfactory	80	09.3
Employees wearing glasses needing changing and who changed to new correction	105	12.3
Employees not wearing glasses whc needed and ordered them	132	15.5
Employees wearing glasses which needed changing but who did not have them changed	14	01.6
Employees with poor vision and strain, not having glasses and who did not obtain them	12	01.4
Total	854	100.0
Employees with normal or maximum vision possible after eye survey	828	96.7
Employees with normal vision before survey	591	69.1
Employees increased to normal or maximum vision by survey	237	27.6

The company sponsored this survey and bore all the general expense in connection therewith. All employees were required to be examined, and the privilege of having the eyes of the members of their families examined was extended to them. All examinations were free to the employees and their families.

Purchase of glasses was optional with the employee. When desired, glasses were sold by the optical company direct to the employee at reasonable prices.

Many unsolicited favorable comments were made by the employees after the survey had been made and glasses procured.

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Table 1.- Accident Data, No. 8 Mine, 1928

PERIOD	Major accidents[1]				Minor accidents			First-aid cases[2]		Calculated data						
	Total nonfatal	Total fatal	Total major accidents	Total days lost	Total number	Total days lost	Total treatments	Total number	Total treatments	Total days lost	Total shifts worked	Days lost per 1,000 shifts worked	Days lost per 1,000 shifts worked to date	Days lost per 1,000 shifts worked last year to date	Accident severity rate[3]	Accident frequency rate[4]
January	0.00	0.00	0.00	0.00	0.00	0.00	0.00	5.00	9.00	0.00	1503.0	0.0	0.0	0.0	0.00	00.0
February	0.00	0.00	0.00	0.00	0.00	0.00	0.00	4.00	16.00	0.00	1495.0	0.0	0.0	0.0	0.00	00.0
March	0.00	0.00	0.00	0.00	0.00	0.00	0.00	5.00	47.00	0.00	1534.0	0.0	0.0	0.0	0.00	00.0
April	0.00	0.00	0.00	0.00	0.00	0.00	0.00	3.00	15.00	0.00	1453.0	0.0	0.0	0.0	0.00	00.0
May	0.00	0.00	0.00	0.00	1.00	6.00	0.00	4.00	24.00	6.00	1650.0	3.6	0.8	0.0	0.45	75.7
June	2.00	0.00	2.00	28.00	0.00	0.00	4.00	2.00	29.00	28.00	1459.0	19.2	3.7	0.0	2.39	171.3
July	0.00	0.00	0.00	60.00	1.00	6.00	20.00	5.00	19.00	66.00	1331.0	49.6	9.6	0.0	6.25	93.9
August	0.00	0.00	0.00	12.00	0.00	0.00	1.00	4.00	7.00	12.00	1536.0	7.8	9.4	0.0	0.98	00.0
September	0.00	0.00	0.00	00.00	0.00	0.00	0.00	7.00	51.00	0.00	1341.0	0.0	8.4	1.6	0.00	00.0
October	0.00	0.00	0.00	00.00	0.00	0.00	0.00	7.00	61,00	0.00	1576.0	0.0	7.5	1.3	0.00	00.0
November	0.00	0.00	0.00	00.00	0.00	0.00	1.00	6.00	64.00	0.00	1488.0	0.0	6.8	1.2	0.00	00.0
December	0.00	0.00	0.00	00.00	0.00	0.00	0.00	2.00	6.00	0.00	1396.0	0.0	6.3	1.0	0.00	00.0
Total, 1928	2.00	0.00	2.00	100.00	2.00	12.00	26.00	54.00	348.00	112.00	17762.0	6.3	6.3	1.0	.79	28.1
Average per month	- -	- -	.16	- -	.16	- -	- -	4.50	- -	- -	- -	- -	- -	- -	- -	- -
Average per accident	- -	- -	- -	50.00	- -	6.00	13.00	- -	6.44	- -	- -	- -	- -	- -	- -	- -

1 Over 13 days lost time.
2 No lost time. Sunday and holidays included effective 7/1/27.
3 Number of days lost per 1,000 hours worked; however, for purpose of this tabulation fatal accidents are not penalized according to scale of the International Association of Industrial Accident Boards and Commissions.
4 Number of lost-time accidents per 1,000,000 hours worked.

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Table 2.- Underground Mining Costs, No. 8 Mine, 1928 [1]

	Operating labor	Repair labor	Total labor	Repair material	Supplies	Total material	Total labor and material	Power	Explosives	Total operating cost per ton ore
Breaking	$0.1319	$0.0026	$0.1345	$0.0053	$0.0154	$0.0207	$0.1552	$0.0092	$0.0721	$0.2365
Loading	.2307	.0029	.2336	.0025	.0094	.0119	.2455	.0009	- - -	.2464
Tramming	.1073	.0233	.1306	.0129	.0203	.0332	.1638	.0138	- - -	.1776
Hoisting	.0313	.0068	.0381	.0025	.0002	.0027	.0408	.0037	- - -	.0445
Pumping	.0012	.0073	.0085	.0035	.0002	.0037	.0122	.0183	- - -	.0305
Mine supervision	.0304	- - -	.0304	- - -	- - -	- - -	.0304	- - -	- - -	.0304
Mine maintenance	.0593	.0067	.0660	.0033	.0053	.0086	.0746	- - -	- - -	.0746
Total ..	0.5921	0.0496	0.6417	0.300	0.0508	0.0808	0.7225	0.0459	0.0721	0.8405

1 Costs cover 168,089 tons of ore mined by open stope and pillar method; development costs are not included in tabulation of costs per ton of ore.

Note: During this period 964 linear feet of 7'-0" x 8'-0" drift were cut at a total cost of $7.87 per foot, divided as follows:

	Amount	Cost per foot
Operating and repair labor	$4,714.66	$4.89
Material and supplies including explosives	1,785.40	1.85
Power	1,089.29	1.13
Total	$7,589.35	$7.87

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Table 3.- Summary of Underground Costs in Units of Labor, Mine No. 8, 1928

(Man hours per ton of rock)

Breaking and development (drifting)	0.122
Loading	0.319
Haulage and hoisting	0.264
Pumping	0.014
Supervision	0.049
Mine maintenance	0.107
Total labor, underground	0.875
Average tons per man per shift	9.14
Labor, percentage of total cost	76.3
Power consumption, kw.h. per ton	4.56

Table 4.- Powder Costs - All Operations

Pounds explosives used per ton rock,	mining only	0.5650
Do.	development	3.2650
Do.	all operations	0.6340
Powder cost per stick,	mining only	$0.0423
Do.	development	0.0556
Do.	all operations	0.0437
Powder cost per ton, all rock broken		0.0827

* * * * * *

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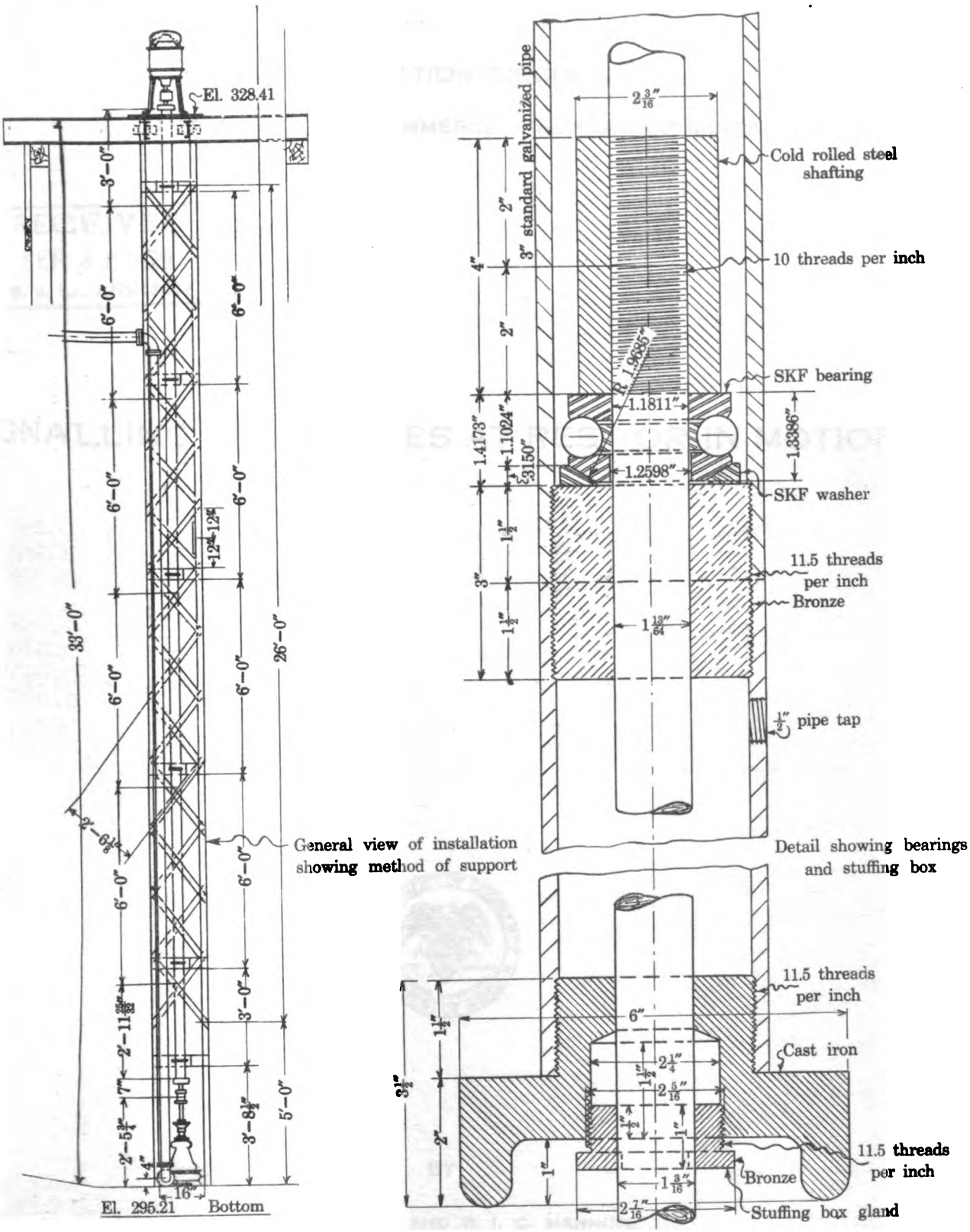

Figure 15. – Shaft pump installation

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